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MILLING METHODS AND COSTS AT THE COPPER QUEEN CONCENTRATOR OF THE PHELPS DODGE CORPORATION, BISBEE, ARIZ.

BY

E. WHITTENAU AND W. B. CRAMER

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MILLING METHODS AND COSTS AT THE COPPER QUEEN CONCENTRATOR OF THE PHELPS DODGE CORPORATION, BISBEE, ARIZ.[1]

By E. Wittenau[2] and W. B. Cramer[3]

INTRODUCTION

This paper describing the concentrator practice of the Copper Queen Branch of the Phelps Dodge Corporation at Bisbee, Ariz., is one of a series of papers on milling methods and costs being prepared by the United States Bureau of Mines.

The Copper Queen Concentrator treats 4,000 tons of copper ore daily by preferential flotation, whereby the copper sulphides are floated and considerable iron pyrite is depressed. During 1929 the heads to the concentrator averaged 2.15 per cent of copper, the recovery was 91.32 per cent of the total copper, and the concentrates produced contained 12.92 per cent of copper.

ACKNOWLEDGMENTS

In the preparation of this paper, credit is due P. L. Schaefer, metallurgist at the Copper Queen concentrator, for much of the assembly thereof and for the preparation of all charts and tables used in connection therewith.

T. G. Chapman, one of the consulting engineers of the U. S. Bureau of Mines, rendered assistance in the arrangement of the material in the paper.

LOCATION OF CONCENTRATOR AND GENERAL DESCRIPTION OF MINE AND MILL

The Copper Queen Concentrator is located 3 miles south of Bisbee, Ariz., on a gently sloping hillside. This location, on one of the southern-most slopes of the Mule Mountains, lends itself admirably to mill construction as it provides an adequate fall for gravity flow of pulp through the mill and out to the tailings ponds on the alluvial valley floor below. Haulage and transportation are easily effected without excessive grades either for the plant railway or for the public carriers.

During the seven years history of the concentrator the ore source has gone through a conversion from an open-pit mine to underground mines. All mining operations have so far been conducted from a central location about 1 mile south of Bisbee and the same plant railway has been used throughout the whole period. The West ore body was extensively mined by

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6404."

2 - One of the consulting engineers, U. S. Bureau of Mines, and concentrator superintendent, Copper Queen Branch, Phelps Dodge Corporation.

3 - One of the consulting engineers, U. S. Bureau of Mines, and consulting engineer, Phelps Dodge Corporation.

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open-pit methods for five and a half years when the depth of the pit recommended a conversion to underground mining methods. This West ore body, known as Sacramento Hill, still continues to produce ore through glory holes in the old pit bottom; ore from this source augmented by other underground ores from the East and Southeast Extension ore bodies makes up the entire tonnage of milling ore. Ore from the glory holes is drawn to bombing chambers on the 400 level of the mine where it is blasted to pass 20-inch grizzlies, after which it passes to underground haulage and hoisting. East and Southeast Extension ores are mined by block caving. All of these milling ores are hoisted at the Sacramento shaft.

Ore haulage to the mill is conducted with trains of six or ten 40-ton capacity side air dump cars which are hauled by locomotive a distance of 2 miles to the crushing plant well up on the hillside.

Figure 1 shows a ground plan of the concentrating plant. The mill buildings are of steel and concrete with galvanized iron sides and asbestos roofs. Concrete is also used extensively for launders, tanks, and machine supports. Overhead cranes have access to all machines, and the craneways extend to the machine and repair shops located in the mill building at one side of the grinding units. The coarse crushing plant is in a separate building located above and at one side of the concentrator building so that crushed ore is delivered sidewise into the fine ore bins at the head of the concentrator.

The concentrator proper comprises four individual and entirely independent units. Each unit includes two primary and two secondary grinding mills, together with classifying, concentrating, and accessory apparatus. The thickening department, with tanks for concentrates and tailings, is placed out in the open in front of and to the side of the concentrator building, and slightly lower on the hill. In front of the tanks and still slightly lower is the filter building, which is directly over the railroad tracks. The mill office, carpenter shop, and warehouse surround the filter plant on approximately the same level.

Compactness has been achieved in this plant and yet free access to the various machines has been retained on all sides with ample room for extensive remodeling from time to time as improved metallurgical and grinding practices dictate. These advantages have been gained by careful planning of launders, pipes, machines and their grouping and by the absence of shafting and belting due to the almost universal use of individual motor drives.

Capacity

As originally designed for coarse grinding, the plant had a daily capacity of 4,500 tons. However, with later tendencies toward finer grinding and the final adoption of grinding which specified that 75.0 per cent of the flotation feed was to pass a 200-mesh screen, the tonnage rate was reduced to 4,000 tons per day. Present tests, with internal unit remodeling, are expected to increase the tonnage rate to 6,000 tons per day.

Water Supply

Ample water for concentrating purposes came as a direct result of a program of water development for this plant. The water source is 1,800 feet underground in a shaft located one-half mile away and over a hill from the concentrator.

Power Supply

Power is derived from generators driven by large Nordberg, FH4-type, Diesel engines

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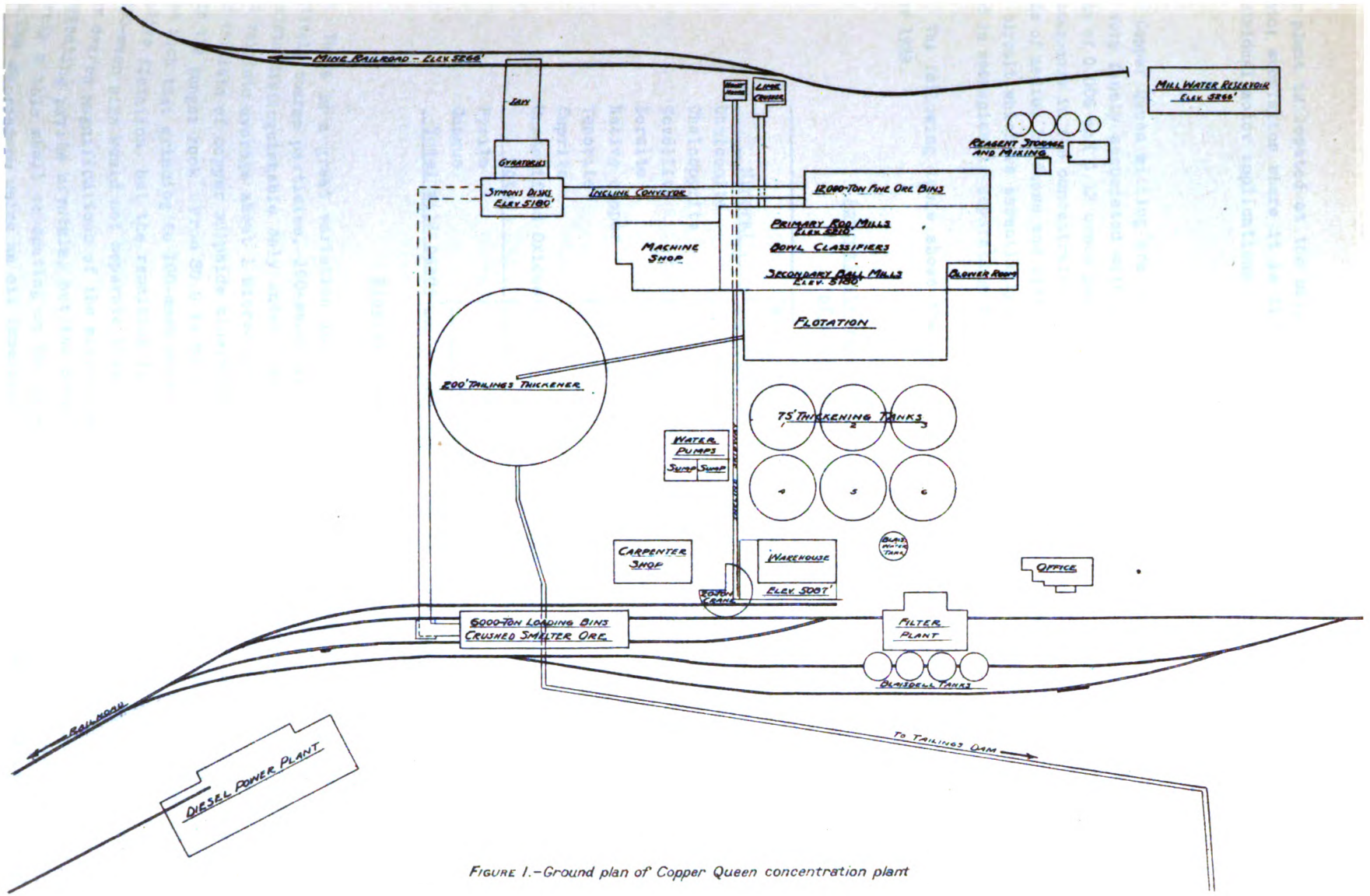

FIGURE 1.—Ground plan of Copper Queen concentration plant

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The power plant is located at the mill site, and power is delivered at 2,200 volts to the concentrator substation where it is distributed at 2,200 and lower voltages for all three-phase individual motor applications.

ORE TREATED

Copper Queen milling ore is a porphyry ore carrying copper mainly as chalcocite, which is very finely associated with pyrite, in a siliceous gangue. Gold and silver occur in amounts of 0.006 and 0.12 ounce per ton respectively, hence they are not of any metallurgical consequence in the concentrator operations. No other valuable metals occur in the ore. The ore is of medium hardness and offers little resistance to crushing, but in the secondary grinding circuit which is essentially a mineral grind, considerable difficulty has been encountered in mechanically separating the copper and iron minerals.

The following table shows the approximate mineralogical composition of the mill heads for 1929.

Approximate mineralogical composition of mill heads, 1929

Mineral	Weight of mineral, per cent	Weight of copper, per cent
Chalcocite	2.32	1.87
Chalcopyrite	0.07	0.03
Covellite	0.13	0.09
Bornite	0.03	0.03
Native copper	Trace	Trace
Tenorite	0.02	0.01
Cuprite	0.02	0.02
Unspecified Oxides	0.13	0.10
Total	2.72	2.15
Pyrite	19.11	
Gangue	78.17	
Total mill heads	100.00	2.15

Mineral Characteristics

There is a great variation in the sizes of the sulphide minerals, ranging from comparatively coarse particles, 100-mesh size or greater, down to extremely fine particles and veinlets distinguishable only under high degree magnifications of the microscope. The smallest veinlets average about 1 micron (0.001 millimeter) in width. The very small particles or veinlets of copper sulphide minerals are practically always in a mass of pyrite and seldom in the gangue rock. From 80.0 to 90.0 per cent of the sulphide copper minerals are in particles such that grinding to 100-mesh size renders the copper and iron minerals separable by selective flotation, but the remaining 10.0 or 20.0 per cent are of such size that grinding to 300-mesh size would not separate them from attached particles. A cursory examination under low degree magnifications of the microscope shows many chalcocite particles cubical in form, suggesting pyrite crystals, but the breaking of these particles shows the chalcocite to be merely a thin shell or coating on the exterior of the pyrite grains. Direct observation with the microscope using an oil immersion lense and obtaining a magnification of 1,500 diameters, shows that a considerable proportion of the chalcocite occurs in an almost unbelievably fine network of interlacing veinlets through the pyrite grains.

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Figure 2 shows a large central grain of pyrite that has been shattered, and in the cracks produced, chalcocite has been deposited. These chalcocite stringers are very minute, some of them about one one-thousandth of a millimeter in width.

Gangue Content

In order to show the gangue content of concentrator heads with respect to the copper minerals, the following analysis representing concentrator heads for 1929 is presented.

Analysis of concentrator heads, 1929.

	Per cent	Ounces per ton
Total copper	2.15	-
Acid soluble copper	0.13	-
Silica	61.6	-
Alumina	10.9	-
Iron	8.9	-
Lime	0.9	-
Sulphur	10.1	-
Gold	-	0.006
Silver	-	0.12
Unaccounted for	5.32	-
Total	100.00	-

None of the gangue minerals, other than pyrite, cause any metallurgical interference in concentrating operations.

Moisture and Slime

Since the conversion from open-pit to underground mining methods the per cent of moisture has increased materially, as has also the percentage of fine material delivered to the crushing plant. Open-pit steam-shovel operations produced and handled lumps of ore so that the run of mine maximum size ranged from 4.0 to 6.0 feet in one dimension. Grizzlies in the mine are now set at 20.0 inches so that the fines content has materially increased by finer reduction of ore at the mine. At the same time the moisture content has increased from 2.50 per cent in 1924 to 3.16 per cent in 1929.

Although Copper Queen ores oxidize rapidly and are very amenable to leaching, little oxidation takes place between mining and milling of the ore because of the short interval of elapsed time between these two operations. Of the oxide minerals present, microscopic examinations have shown that chalcanthite and other sulphates are predominant and occur as films and dust over the sulphide minerals. These lighter, finer oxides naturally follow the slimes in milling operations and account for approximately 10 per cent of the total copper in the slimes as fed to the slime flotation section.

BRIEF HISTORY OF CONCENTRATOR OPERATIONS

The plant was designed by H. Kenyon Burch and is based on pilot plant operations from 1917 to 1919 in which combined gravity and flotation and all flotation methods were used. The design adopted called for the treatment of the ore by both gravity and flotation

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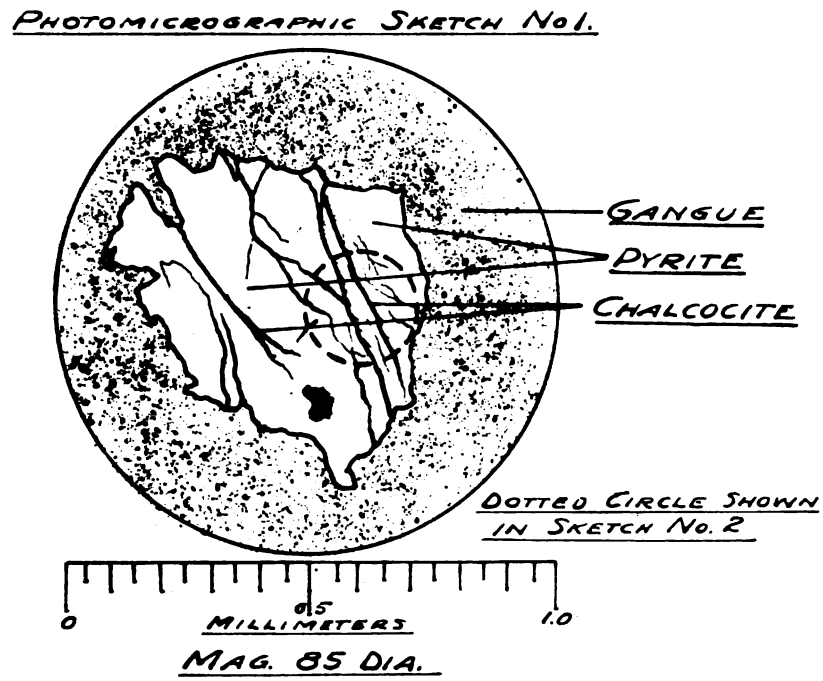

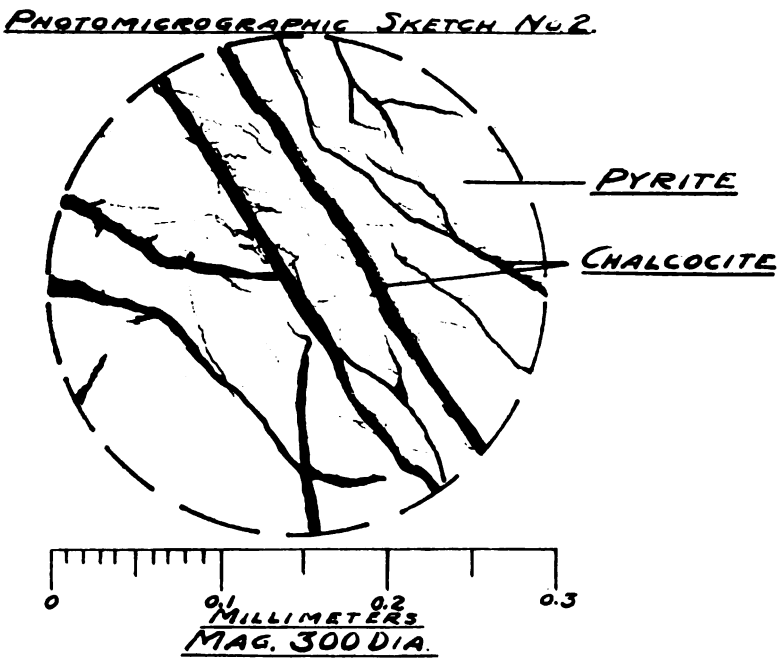

FIGURE 2.—Sketches of microsections of Copper Queen milling ore

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methods, with flotation constituting an intermediate step between roughing and finishing tables. This method of treatment was chosen because the characteristics of the ore made fine grinding unnecessary for the purpose of separating gangue from sulphide minerals, and also because the requirements for smelting operations called for a large proportion of coarse concentrates and pyrite for the fluxing of siliceous ores. Approximately 50.0 per cent of the total concentrates was produced on 40 roughing tables treating the primary rod-mill discharge. These concentrates averaged between 4.0 and 5.0 per cent of copper, 25.0 per cent of insoluble, and about 30.0 per cent of iron. The tailings from these tables were deslimed and ground to minus 48-mesh size in the secondary rod mills and treated by flotation in 16 rougher-cleaner cells of the Callow pneumatic type. Flotation produced concentrates of 8.0 per cent of copper, 20.0 per cent of insoluble, and 30.0 per cent of iron; these concentrate sconstituted 48.0 per cent of the total concentrates. Eighty clean-up or finishing tables then completed the concentrating process by making concentrates containing 4.0 per cent of copper, 20.0 per cent of insoluble, and 38.0 per cent of iron.

Concentrating operations started in April, 1923, and early operations indicated the advisability of removing more pyrite and insoluble from the concentrates in view of the constant depletion of siliceous copper ores in the mines. Production of a higher grade of concentrates would result in a decided saving in freight and smelting charges per ton and a reduction in smelter slag losses due to the removal of slag-forming constituents at the mill. In bringing about this condition of producing cleaner concentrates there were involved: The elimination of all gravity concentrating machines; grinding sufficiently fine not only to separate the sulphide minerals from the gangue but to separate the copper and pyrite minerals; improved classification; the adoption and rearrangement of new-type flotation apparatus; and changes in the character of flotation reagents. A marked reduction in the amount of equipment and changes in the character of equipment accompanied the change in metallurgical treatment, and there are probably best shown by the following comparison of the original and present grinding and concentrating equipment.

Comparison of original and present grinding and concentrating equipment

Original equipment of 1923			Equipment in present use		
Number	Size	Machines	Number	Size	Machines
8	6 by 12-foot	Primary rod mills	8	6 by 12-foot	Primary rod mills
4	8 by 16-foot	Desliming classifiers	8	15-foot	Bowl classifiers
40		Roughing tables	8	6 by 12-foot	Secondary ball mills
8	6 by 12-foot	Secondary rod mills	8	6 by 23-foot	Circulating classifiers
8	6 by 23-foot	Circulating classifiers	12		Forrester rougher-cleaner cells
16		Callow rougher cells	4		Concentrates classifiers
16		Callow cleaner cells	1	75-foot	Concentrates thickener
80		Finishing tables	1	14-foot	Filter
4	75-foot	Concentrates thickeners	4		Blaisdell settling tanks
3	14-foot	Filters			
4		Blaisdell settling tanks			

The changes mentioned took place in the following sequence:

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Year	Change adopted
1923	Concentrator started operating. The lime circuit was adopted during the first week of operation.
1924	Finishing tables stopped operating and roughing tables were eliminated, thereby producing all-flotation treatment.
1925	Organic chemical flotation reagents adopted.
1926	Finer grinding adopted.
1927	Forrester flotation machines adopted. Flotation flow sheet revised to return all middlings to and take all final concentrates from the primary flotation cells.
1928	Minor refinements in recently adopted practices.

The following tabulation shows improvements in metallurgical results and direct costs of concentrating since the beginning of operations at this plant.

Comparison of concentration recovery and direct costs since beginning of operations

Year	Direct cost per ton	Copper recovery, per cent	Ratio of concentration
1923	$0.844	89.24	4.79
1924	.703	90.71	4.49
1925	.653	87.97	5.95
1926	.568	86.93	6.35
1927	.597	87.53	9.21
1928	.531	91.26	7.93
1929	.493	91.32	6.58

Concentrates grade and attendant ratios of concentration were slightly lowered during the past year owing to the change in source of an increasing tonnage of mill ore. 1928 was the last year that the concentrator treated West ore body or Sacramento Hill ore exclusively. Some ores of the newer group are quite refractory, and with present metallurgical knowledge they present a definite problem in concentrates grade.

One of the four concentrator units is at present operating under test conditions with circulating classifiers in closed circuit and the primary rod mill and 16-foot circulating classifiers in closed circuit with 8 1/2 by 12 foot secondary ball mills. This test work was so planned that the first item—that of primary classification—was determined before the installation of the larger secondary grinding equipment. Thus the added advantage of primary classifiers on a normal unit was first determined, and now the test is to be continued to determine the advantages of the enlarged secondary unit. It may be stated this grinding test work is not being conducted to achieve finer grinding but is for the sole purpose of adding capacity by internal remodeling. It has been proved that the present unit tonnage rate of 1,000 tons per day can be increased 10.0 per cent by the use of primary classifiers alone. The balance of the test is as yet uncompleted.

PRESENT METHOD OF CONCENTRATING

Coarse Crushing

Figure 3 gives the flow sheet of the coarse crushing plant and Table 1 gives rated

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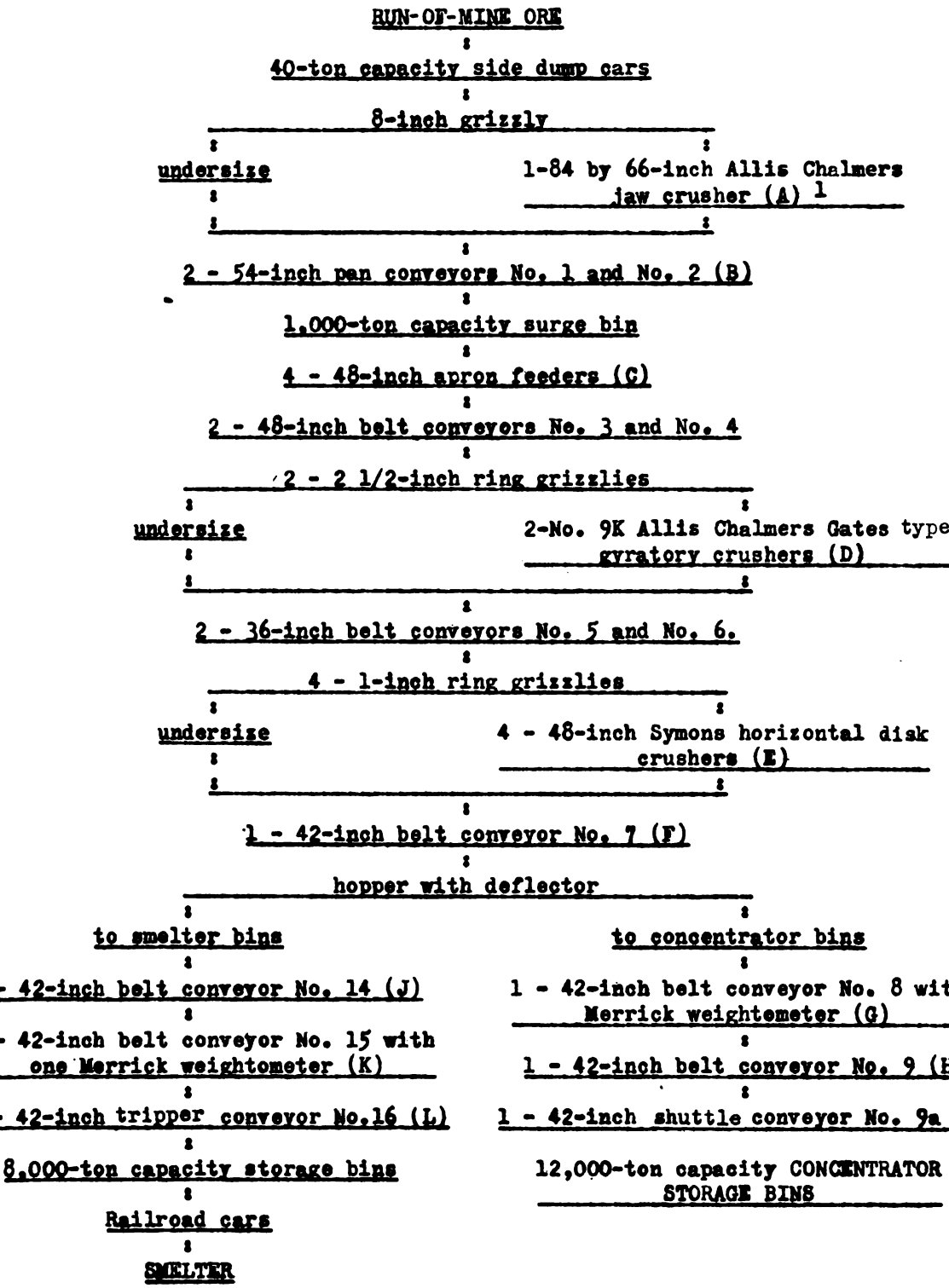
RUN-OF-MINE ORE
40-ton capacity side dump cars
8-inch grizzly
undersize
1-84 by 66-inch Allis Chalmers jaw crusher (A) 1
2 - 54-inch pan conveyors No. 1 and No. 2 (B)
1,000-ton capacity surge bin
4 - 48-inch apron feeders (C)
2 - 48-inch belt conveyors No. 3 and No. 4
2 - 2 1/2-inch ring grizzlies
undersize
2-No. 9K Allis Chalmers Gates type gyratory crushers (D)
2 - 36-inch belt conveyors No. 5 and No. 6.
4 - 1-inch ring grizzlies
undersize
4 - 48-inch Symons horizontal disk crushers (E)
1 - 42-inch belt conveyor No. 7 (F)
hopper with deflector
to smelter bins
1 - 42-inch belt conveyor No. 14 (J)
1 - 42-inch belt conveyor No. 15 with one Merrick weightometer (K)
1 - 42-inch tripper conveyor No.16 (L)
8,000-ton capacity storage bins
Railroad cars
SMELTER
to concentrator bins
1 - 42-inch belt conveyor No. 8 with Merrick weightometer (G)
1 - 42-inch belt conveyor No. 9 (H)
1 - 42-inch shuttle conveyor No. 9a (I)
12,000-ton capacity CONCENTRATOR STORAGE BINS

[1] Capital letters in parenthesis refer to motor horsepower given in Table 1.

Figure 3.- Flow sheet of coarse crushing plant

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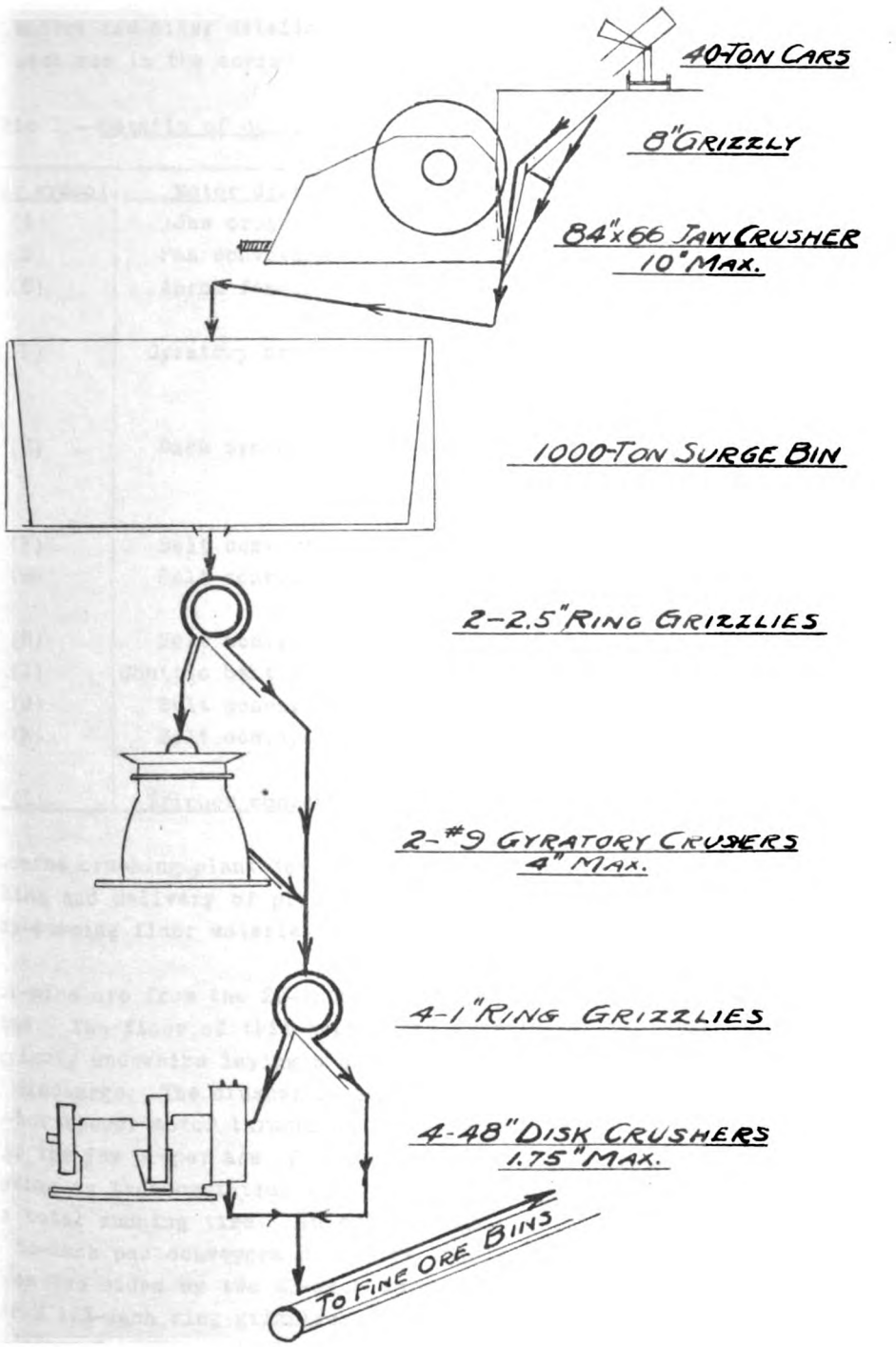

FIGURE 4.-Diagrammatic arrangement of machines in the coarse crushing plant

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horsepowers of motors and other details of the crushing plant. Figure 4 shows a diagramatic arrangement of machines in the coarse crushing plant.

Table 1.—Details of coarse crushing plant as designated in Figure 3

Letter symbol	Motor driving	Horsepower	Remarks
(A)	Jaw crusher	250.0	English-type rope drive.
(B)	Pan conveyors	75.0	Through triple reduction
(C)	Apron feeders	5.0	One motor for each feeder.
(D)	Gyratory crushers	125.0	One motor drives one crusher, grizzly and feeder belt.
(E)	Disk crushers	125.0	One motor drives two crushers, grizzlies, and feeder belt.
(F)	Belt conveyor	15.0	
(G)	Belt conveyor	100.0	Long incline pull, 320 feet.
(H)	Belt conveyor	25.0	
(I)	Shuttle belt conveyor	15.0	
(J)	Belt conveyor	15.0	
(K)	Belt conveyor	75.0	Long horizontal pull of 497 feet.
(L)	Tripper conveyor	25.0	

The coarse crushing plant has an almost straight ore path through the three stages of coarse crushing and delivery of product to the fine ore bins. Each crushing stage has grizzlies for by-passing finer material.

Run-of-mine ore from the 20-inch grizzlies is dumped into the jaw crusher hopper, 40 tons at a time. The floor of this hopper has grizzly bars to by-pass minus 8-inch size material, the grizzly undersize laying a cushion on the 54-inch pan conveyors which receive the jaw-crusher discharge. The crusher is an 84 by 66 inch Allis Chalmers Blake type machine driven by a 250-horsepower motor through an English-type rope drive of 13 ropes. All liners and jaw plates in the jaw proper are of manganese steel. This machine will handle 2,500 tons per hour, but owing to transportation intervals it is actually crushing approximately 12.0 per cent of the total running time. In order to give a constant feed to the remainder of the plant, the 54-inch pan conveyors discharge into a 1,000-ton capacity surge bin. This bin is drawn from two sides by two 48-inch apron feeders and belt conveyors which deliver the product over 2 1/2-inch ring grizzlies to two No. 9K Gates-type Allis Chalmers gyratory crushers, belt-driven from individual motors. The gyratory crushers discharge a 4-inch product which is carried by two 36-inch conveyors over 1-inch ring grizzlies to four 48-inch Symons horizontal disk crushers which are belt-driven in pairs. The Symons crusher discharge products showing a maximum size of 1.75-inch, with 13.5 per cent through 200-mesh, is the final crushing plant product and as such is delivered to the 12,000-ton capacity fine ore bins by a 42-inch belt over a Merrick weightometer and through a shuttle conveyor.

Closed-circuit trommels or screens are not operated in this plant, consequently no accurate screenings are conducted on the various intermediate stages of coarse crushing. Table 5 gives a screen analysis of the final product of the coarse crushing plant.

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That portion of the coarse crushing plant from the surge bins on down, which includes the gyratory and Symons crushers, has an hourly capacity of 400 tons - ample capacity to supply enough crushed ore for continuous operation of the concentrating plant by working two shifts six days a week.

General practice favors finer feed for the grinding mills, but Copper Queen ore bodies unfortunately lie in the drainage region of a considerable watershed. During the rainy season much difficulty is encountered in crushing even to the coarse sizes noted, and production would be practically nil if finer sizes were attempted during the wet periods.

Concentrator

The concentrator proper consists of four units and treated an average of 4,281 tons of ore per 24 hours of operating time during 1929. The maximum tonnage treated during 1929 amounted to 4,900 tons per 24 hours.

Figure 5 is a flow sheet of the concentrator and Table 2 gives rated horsepower of motors and other details of the concentrator. In order to assist in clarifying the flow sheet of Figure 5, a longitudinal section of the concentrator is given in Figure 6. The successive steps of concentrating consist of primary grinding, bowl classification, secondary grinding in closed circuit, flotation, thickening, and filtering of flotation concentrates.

Table 2.--Details of concentrator as designated in Figure 5

Letter symbol	Motor driving	Horsepower	Remarks
(M)	Belt conveyor	10.0	Drives two conveyors, No. 10 and No. 11.
(N)	Belt conveyor	5.0	
(O)	Primary mills	150.0	Through Falk reduction gears.
(P)	Bowl classifiers	7.5	Through Palmer-Bee reduction gears.
(Q)	Secondary mills	150.0	Through Falk reduction gears.
(R)	Circulating classifiers	15.0	Drives two classifiers.
(S)	Concentrates classifiers	1.0	Through back gears,
(T)	200-foot thickener	5.0	
(U)	75-foot thickeners	3.0	
(V)	75-foot thickeners	3.0	
(W)	Oliver filters	3.0	
(X)	Flotation blowers	125.0	At low pressure of 1 5/8 pounds.

Primary Grinding

Figure 7 gives a diagramatic flow sheet of primary and secondary grinding of one unit of the concentrator.

A system of six apron feeders with one collecting and two elevating conveyor belts brings the ore from the fine ore bins over a Merrick weightometer to the two primary rod mills of each unit. The primary rod mills are 6 by 12 feet in size, of Marcy design, and

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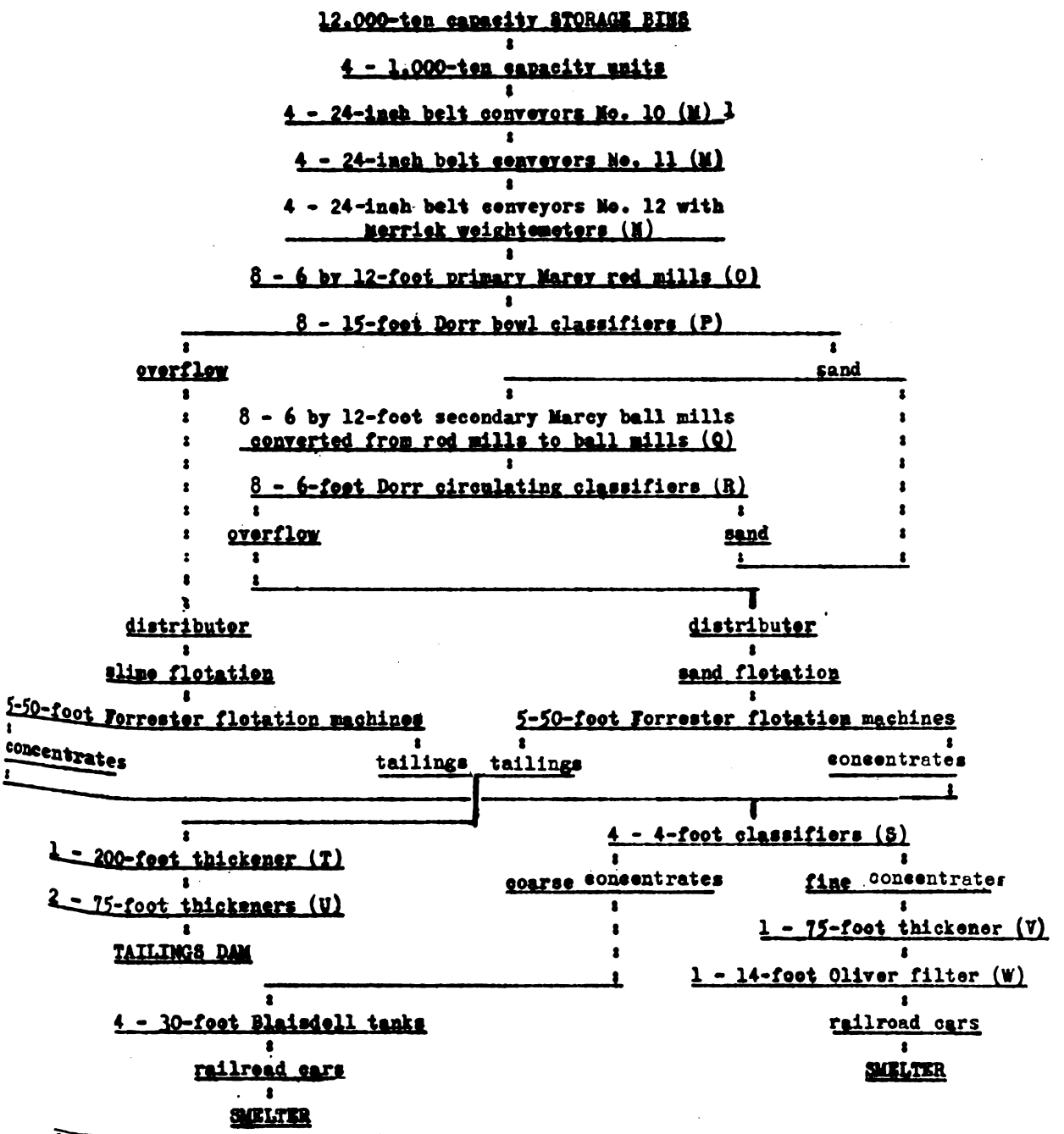
12,000-ton capacity STORAGE BINS
4 - 1,000-ton capacity units
4 - 24-inch belt conveyors No. 10 (M) 1
4 - 24-inch belt conveyors No. 11 (M)
4 - 24-inch belt conveyors No. 12 with Merrick weightometers (M)
8 - 6 by 12-foot primary Marcy rod mills (O)
8 - 15-foot Dorr bowl classifiers (P)
overflow
sand
8 - 6 by 12-foot secondary Marcy ball mills converted from rod mills to ball mills (Q)
8 - 6-foot Dorr circulating classifiers (R)
overflow
sand
distributor
distributor
slime flotation
sand flotation
5-50-foot Forrester flotation machines
5-50-foot Forrester flotation machines
concentrates
tailings
tailings
concentrates
4 - 4-foot classifiers (S)
1 - 200-foot thickener (T)
2 - 75-foot thickeners (U)
coarse concentrates
fine concentrates
TAILINGS DAM
1 - 75-foot thickener (V)
1 - 14-foot Oliver filter (W)
4 - 30-foot Blaisdell tanks
railroad cars
railroad cars
SMELTER
SMELTER

1 Capital letters in parenthesis refer to meter horsepower given in Table 2.

Figure 5.- Flow sheet of concentrator proper

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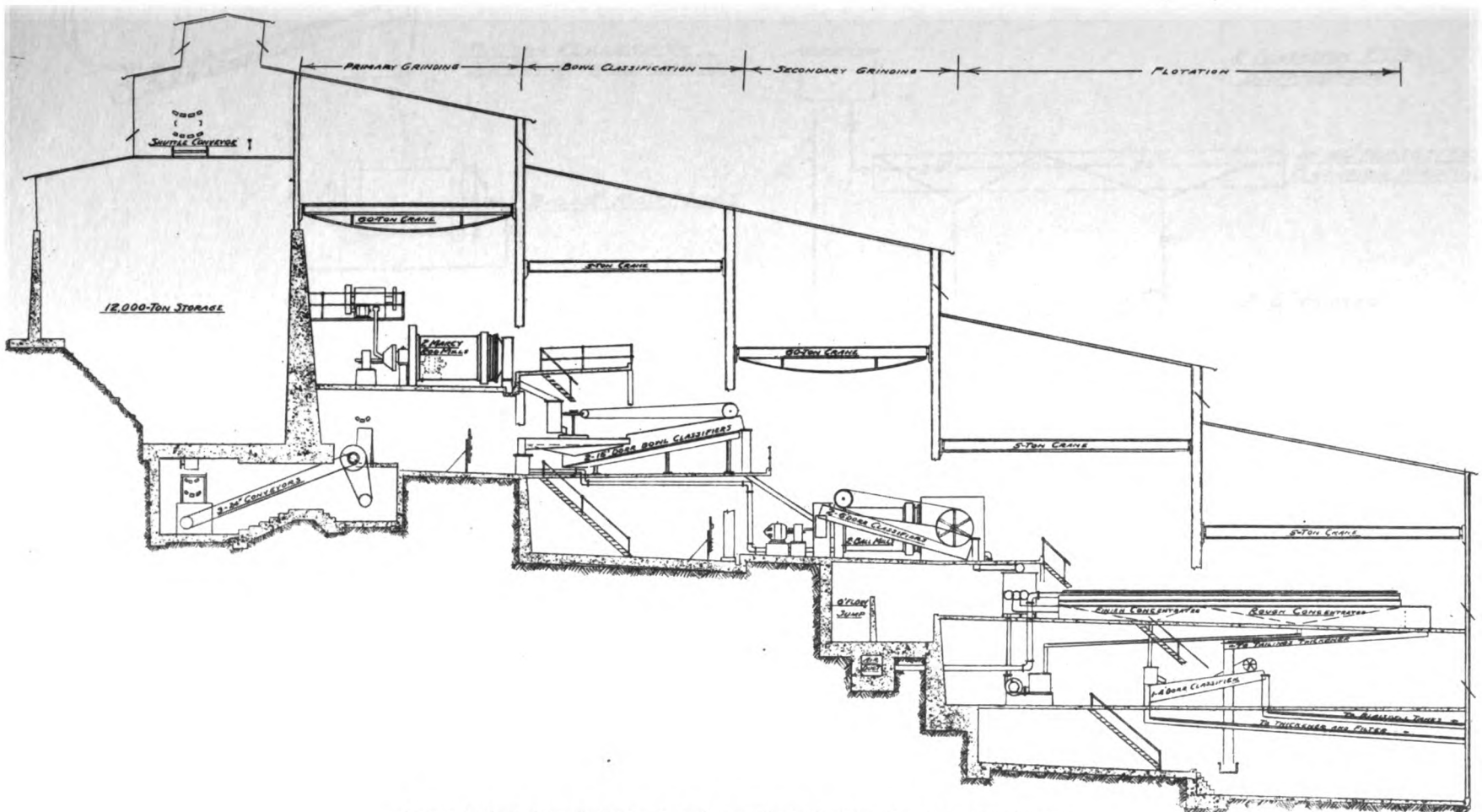

FIGURE 6.—Typical longitudinal section of concentrator building, showing unit equipment

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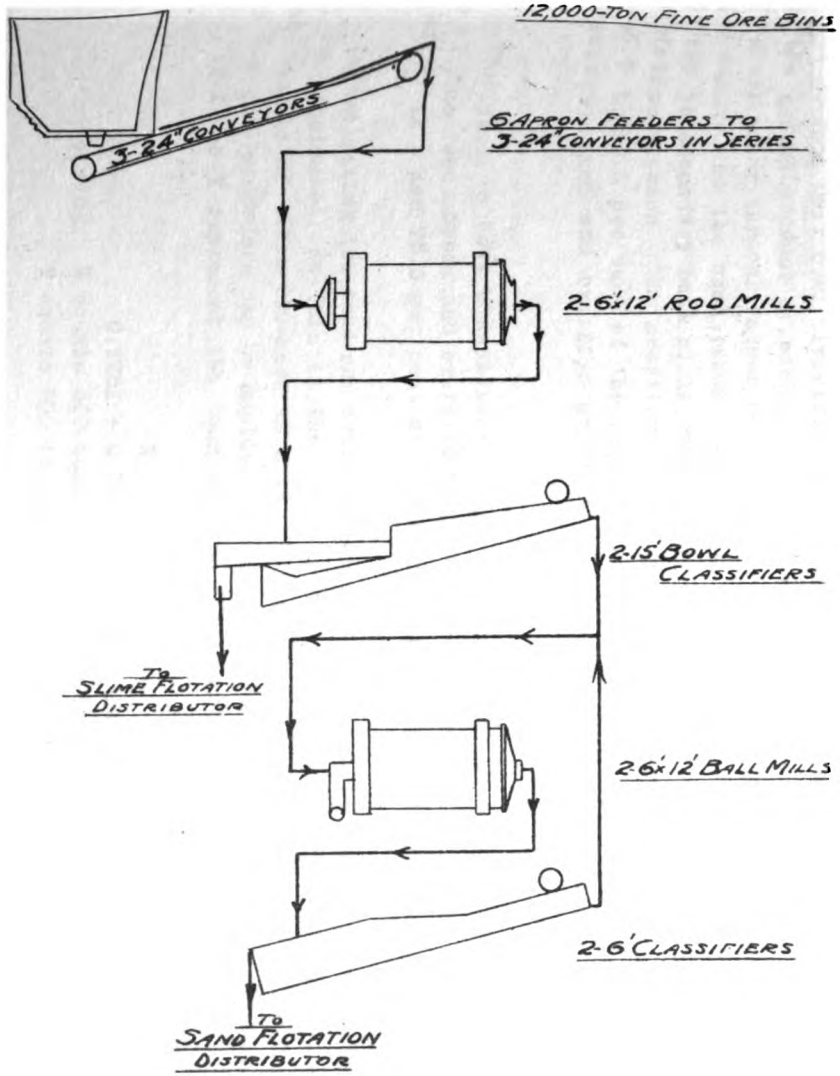

FIGURE 7.—Diagrammatic flow sheet of primary and secondary grinding of one concentrator unit

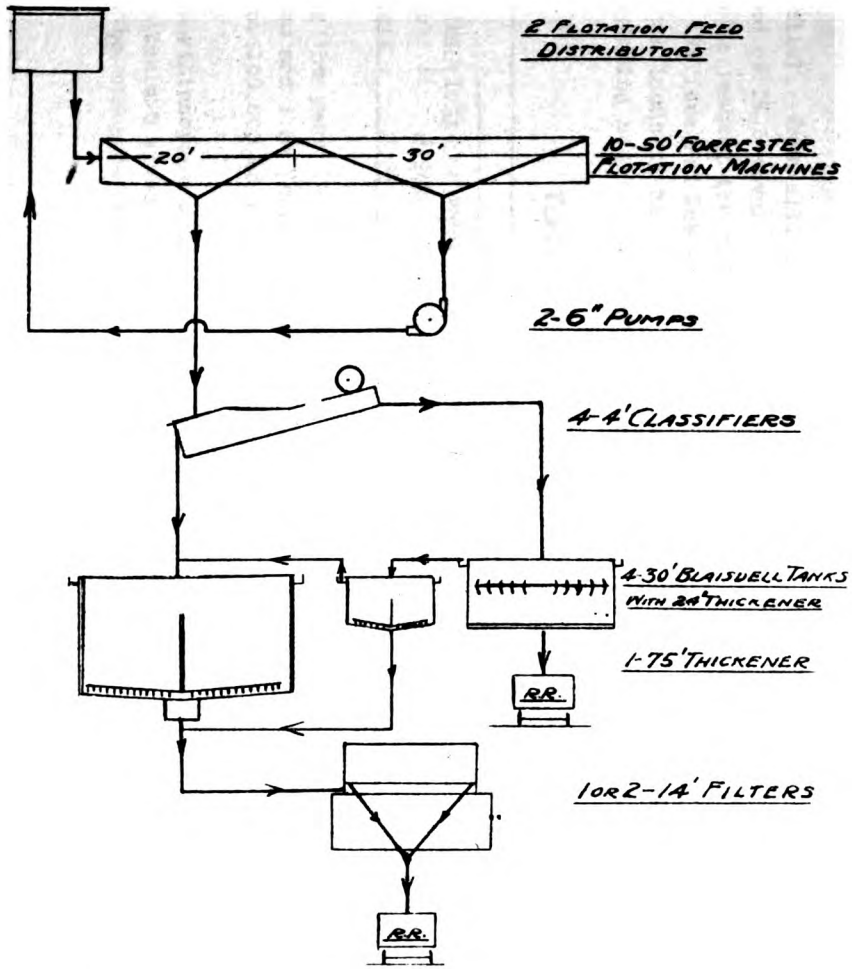

FIGURE 8.—Flow sheet of flotation treatment and concentrates handling

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operate in parallel. Each mill is driven by a 150-horsepower motor through Falk reduction gears at a speed of 17.5 revolutions per minute. Dry crushed lime is automatically fed to the conveyor belts feeding the rod mills. Three-inch, high-carbon, steel rods are used for grinding media. Various rod loads have been tried in these mills ranging from 12 to 19 tons, but for given conditions it has been found that the limit load of 19 tons is the most efficient mass of grinding media. Typical analyses of rods and liners are as follows:

Typical analyses of rods and liners

	Per cent					Brinell
Item	Total carbon	Silicon	Sulphur	Manganese	Phosphorous	hardness
Rods	0.960	0.194	0.039	0.813	0.029	295
Liners	2.68	1.220	0.177	0.269	0.457	252

During the year 1929 these primary mills used 0.45 pounds of liners per ton of primary mill feed and 1.34 pounds of rods per ton of ore ground. Liner life in the primary mills ranges from 100,000 to 110,000 tons of ore ground.

The consistency of pulp in the rod mills is maintained between 73.0 and 75.0 per cent of solids. Table 5 gives a screen analysis of the rod-mill discharge product and shows the material to be minus 14 mesh with approximately 45 per cent of the product minus 200 mesh.

Bowl Classification

A 15-foot Dorr bowl classifier with a 6 by 28 foot rake compartment is provided to classify the ground product of each primary rod mill. Each bowl classifier is driven by a 7.5-horsepower motor through Palmer-Bee reducers, the bowl rakes having a speed of 5 revolutions per minute and the sand rakes operating at 19 strokes per minute. The sand product is delivered to secondary ball mills, and the overflow product constitutes the feed for the slime flotation section. The overflow product contains 22.0 per cent of solids and amounts to from 45.0 to 50.0 per cent of the concentrator feed tonnage. The relative proportion of bowl classifier sand and overflow products are determined periodically by the following method.

The feed to bowl classifiers of one unit is given as 1,000 tons and the screen analyses of the feed, sand, and overflow products are assumed, for purpose of illustration, to show 44.0, 12.5 and 75.5 per cent of minus 200-mesh material, respectively.

In estimating the sand and overflow tonnages by this method only the minus 200-mesh material is considered, as this is the limiting and defining size at this plant and also since the tonnage of minus 200-mesh material in the ground product is large. For absolute accuracy the same procedure can be applied to all sizes of the concentrator screen analyses.

If X and Y represent the tons of sand and overflow products respectively, then:

$$X + Y = 1{,}000$$

$$0.125X + 0.755Y = 0.44 \times 1{,}000$$

Solving: X equals 500 tons or 50 per cent as sand product.
Y equals 500 tons or 50 per cent as overflow product.

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As contracted this procedure may be stated as:

$$\text{Per cent of sand tonnage} \quad \frac{100(O - F)}{(O - S)}$$

where O per cent of minus 200-mesh material in the overflow product.
F per cent of minus 200-mesh material in the feed to the classifier.
S per cent of minus 200-mesh material in the sand product.

Classifier efficiency is also developed on an absolute standard by employing the following well-known formula.

$$\text{Per cent recovery} \quad \frac{(H - T) \times C}{(C - T) \times H},$$

where H per cent of metal in heads,
C per cent of metal in concentrates,
T per cent of metal in tailings.

In using this formula for computing classifier efficiency, per cent weight of the determining screen size, which for the Copper Queen plant is minus 200-mesh material as already stated, is substituted for per cent of metal. The per cent of minus 200-mesh material in the classifier feed is represented by H, the per cent of minus 200-mesh material in the overflow by C, and the per cent of minus 200-mesh material in the sand product by T. The classifier efficiency, as computed at this plant, is therefore the per cent of the total minus 200-mesh material overflowed by the classifier. This formula can be applied to any limiting screen size material representing an adequate tonnage for accuracy or to the entire series of screen products.

Classification in the bowl classifiers has been improved by increasing the speed of the bowl spider rakes. When the original speed of 3.25 revolutions per minute was employed, the result was that the overflow product contained a large percentage of fine material but had a corresponding lesser tonnage. When the speed of the spider rakes was increased to 5.0 revolutions per minute, the overflow contained coarser material up to the limit for flotation feed, 75.0 per cent minus 200-mesh, and there was a consequent gain in overflowed tonnage, thus leaving less tonnage of sand as feed to the secondary ball mills. The general effect of this alteration may be noted in the following tabulation of results from one unit during the test period.

Comparison of results obtained with low and high spider speed

	Low spider speed, 3.25 revolutions per minute	High spider speed, 5.0 revolutions per minute
Minus 200-mesh material in feed, per cent	46.2	45.4
Minus 200-mesh material in sand, per cent	15.5	12.5
Minus 200-mesh material in overflow, per cent	85.3	75.0
Overflow tonnage, per cent of feed	44.0	52.8
Sand tonnage, per cent of feed	56.0	47.2

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When the bowl classifier rakes operated at a speed of 3.25 revolutions per minute, the classifier efficiency amounted to 81.21 per cent and the ratio of tons of classifier feed to tons of overflow product amounted to 2.27. With the higher rake speed, of 5.0 revolutions per minute, the classifier efficiency was increased to 87.03 per cent and the ratio of tons of classifier feed to tons of overflow product was reduced to 1.89.

Backwash water has been tried in the bowl classifiers to clean the sand product fed to the secondary grinding mills. This practice did make a very clean separation of slime from sand and greatly increased the efficiency of the classifier. Operation, however, is not so much concerned with classifier efficiency as with the efficiency of the grinding circuit as a whole. It has been demonstrated with Copper Queen ores that better overall grinding efficiency results by operating the bowl classifiers as described and without backwash water.

Secondary Grinding

The secondary grinding department, for the grinding of bowl classifier sands, continues with the unit idea and consists of two ball mills operating in closed circuit with two drag classifiers, for each unit. These ball mills are converted 6 by 12 foot Marcy rod mills. With the advent of fine-grinding, extensive testing of Copper Queen ores indicated that ball mills were considerably more efficient for closed-circuit grinding. As previously stated, this secondary grinding operation is a distinct mineral-grinding operation for the purpose of separating chalcocite from pyrite.

When converted to ball mills, the mills retained the same speed of 17.5 revolutions per minute through individual 150-horsepower motors and Falk reduction gears. Short-wave liners replaced long-wave liners, high-center discharge replaced low-center discharge, and 3.5-inch chilled semisteel balls replaced 3.0-inch high-carbon steel rods.

The ball-mill pulp is maintained between 70.0 and 80.0 per cent of solids. A typical analysis of liner material is given in the following tabulation.

Typical analysis of linear material

Per cent					Brinell hardness
Total carbon	Silicon	Sulphur	Manganese	Phosphorous	
2.82	1.200	0.185	0.296	0.387	275

Because of the fact that ball mills were not adopted for secondary grinding until 1928, little data is available on the life of ball-mill liners. Two sets have been worn out to date, one of which ground 386,000 and the other 256,000 tons of ore. This tonnage is based on tons of original feed to the primary rod mills.

The liner consumption amounted to 0.08 pounds and the ball consumption amounted to 1.41 pounds per ton of primary rod-mill feed ground during the year 1929.

The ball mills operate in closed circuit with 6 by 23 foot, Model D, Dorr circulating classifiers, the classifier overflow product constituting the feed to the sand flotation machines. Elevating scoops, mounted on a horizontal shaft driven from the classifier head motion, effect adequate elevation for feeding the mill discharge into the classifiers.

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The circulating classifiers are driven from line shafts, one being provided for each unit, and each classifier is made independent of the other by means of a clutch arrangement. Incidentally, the line shafts for the circulating classifiers are the only line shafts in the concentrator building. It may be noted that in the newly remodeled secondary grinding unit, the classifiers have individual motor drives; thus all line shafting in the concentrator building is eliminated.

The overflow product of the circulating classifiers contains from 23.0 to 25.0 per cent of solids. The circulating load is never constant because of the varying feed values; no attempt is made to determine or control this factor as it merely becomes a function of the sulphide mineral content of the ore and changes with every ore change, which in the past has occurred two or three times per shift. Constant attention is directed to the final overflow product with respect to fineness and per cent of solids.

The independent operation of concentrator sections through unit construction has been noted. The operation of a concentrator section can be further divided so that but one half of a section operates at one time, an obvious advantage with respect to the effect of repairs on operating time.

Spare mills, reduction gears, motors, classifier parts, and accessory supplies are always kept on hand so that repairs can be made immediately in any half section. This practice results in a definite time-saving factor. Of the total time lost through delays in 1929 less than 25.0 per cent was due to delays from repairs to the grinding and attendant operations.

Table 5 gives typical screen analyses of the feed to the ball mills and the classifier overflow product.

Flotation

Figure 8 is a flow sheet of flotation treatment and concentrates handling. Flotation treatment is effected in Forrester-type machines. The treatment of the bowl classifier overflow product, which contains the primary slime and slime produced in the primary rod mills, is separate from the treatment of the circulating classifier overflow product, which consists of reground bowl sands. The entire concentrator contains ten 50-foot Forrester machines, five of which treat the bowl classifier overflow product and five treat the circulating classifier overflow product. Flotation treatments in the slime and sand sections are the same and consist in producing finished concentrates for the first 20 feet of each Forrester machine and a middlings product for the remaining 30 feet, the middlings product being returned to the distributor feeding the Forrester machines. Cleaning of flotation rougher concentrates is not practiced.

The Forrester-type flotation machine was introduced at this plant in September, 1926, and several months later replaced the Callow-type machine throughout the plant. Among the determining factors, which contributed to the adoption and success of the Forrester machine at this plant, was its ability of making a lower grade of concentrates, relative to grade of feed, especially when the flotation feed increased in copper content. The condition of intermittent high copper content feed was not abnormal at the Copper Queen plant when the Forrester machine was adopted. When such a condition occurs the flotation machine must be able to produce a relatively lower grade of concentrates in order to maintain a normal recovery of copper. As stated, one of the advantages of this machine is its ability to produce

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a greater bulk of concentrates on short notice. Another advantage of the Forrester machine, when operating on Copper Queen ores, is the fact that violent agitation is achieved in this machine and this condition is apparently essential for best results on these ores.

The Forrester machines, as installed at present, require practically no operation or attention. The operating speed of the machines varies automatically with the character of the ore being treated, and the machines have been found to adjust themselves to variable conditions within the normal limits of operation.

All Forrester machines were built directly into the old Callow tanks, therefore the same outside dimensions and the former uniformity of construction were retained. Since the introduction of the Forrester machine it has been given extensive tests side by side with all modern flotation equipment of both mechanical and pneumatic type, but to date no changes have been effected.

In each flotation machine there are 203 half-inch air pipes spaced on 3-inch centers, thus providing an approximate total orifice discharge area of 40 square inches. The main air header for each machine is 10 inches in diameter and provides sufficient volume to maintain an even pressure throughout the 50 feet of machine length. The ultimate air distribution through half-inch pipes insures a fine and thorough dissemination of air throughout the pulp.

Three No. 8 Roots blowers furnish ample air for the ten 50-foot machines at 1.5 to 1.6 pounds pressure. The power consumed amounted to 1.66 kilowatt hours per ton of flotation feed during the 1929 period. Blower speed was increased from 125 to 150 revolutions per minute to produce the large volume of air at the low pressure required. The blower room is located to one side of the secondary grinding section and delivers air through an air tunnel in the foundation work of the plant. Laterals are taken off for each two machines. No measurement of air volume is made at this plant, as it is sufficient to know the pressure and cost through the power consumed per ton of ore treated.

Lime, pine oil, and xanthate constitute the total list of reagents used at this plant.

Dry crushed lime is fed to the primary rod mill feeder belts in the amount of 5. pounds per ton of ore. About 2.0 pounds of this amount is used to neutralize the acidity of the ore and the balance of 3.0 pounds acts as a depressor of pyritic iron. Because of the high alkalinity carried, the accuracy of pH control is not necessary. Hourly determinations of alkalinity are made with $\frac{N}{50}$ sulphuric acid solution, using phenolthalein as the indicator. Flotation feed pulp normally contains 0.6 to 0.8 pound of free CaO per ton of water.

Steam distilled pine oil is used in the amount of 0.04 pound per ton of ore. Copper Queen ores froth very readily, and this fact coupled with the requirement of high pulp level in the flotation machines makes the pine oil consumption very low. This reagent is fed to the bowl and circulating classifier overflow pulps.

Potassium ethyl xanthate is used in amount of 0.10 pound per ton of ore treated. The xanthate is dissolved outside the mill building and a pipe system brings a 12.0 per cent solution to centrally located feeders in the mill. A main feeder, of the dipper type, is provided for calibration, after which further unknown dilution is effected and the xanthate solution is fed to every classifier overflow pulp through a gravity system.

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Xanthate used in conjunction with lime is an important metallurgical factor at this plant. Low xanthate and high lime make for a relatively high ratio of concentration, while high xanthate and low lime give the reverse effect. Of these reagents, the xanthate variation is the more readily controlled and will give quicker results. Due to the large amount of return water used, which is approximately 85.0 per cent of the total water, a rapid change in lime content is impossible, but if a high ratio of concentration is desired over a long period, lime is gradually increased to the desired amount. If during this period a short run of high-grade ore is encountered which on a high concentration ratio basis would cause excessive losses in the tailings, the xanthate content is increased to overcome the depressor effect of lime.

The flow sheet in present use was arrived at after many variations had been tried with respect to cleaning, recleaning, and regrinding. From these tests and information gained by microscopic analyses, it has been found that with this ore the major factor in the control of concentrates grade is the immediate floating and disposal of all available finished material. Due to the high pyritic content, any subsequent treatment along the lines of recleaning has tended to float additional iron with each successive stage. Pyrite is the only diluent factor in Copper Queen concentrates.

As noted, finished concentrates are made in one operation and the grade is regulated primarily by the control of the xanthate. A subsequant reference will comment on a colorimeter installation for hourly determinations of copper in the feed, the concentrates, and the tailings. If this hourly analysis shows very low-grade concentrates, the amount of xanthate is cut down until the required grade of concentrates is produced, providing this does not seriously affect the tailings losses. The operators are furnished with a schedule showing the grade of concentrates and tailings required from a feed of a given copper content This close control shows a response to change in the collector reagent within 30 minutes after a change is made. The actual contact or conditioning of reagents with the pulp is accomplished in distributor boxes which collect all fine grinding products and redistributes them to the flotation machines. As the xanthate and pine oil are fed to the classifier overflow pulps it is in this collection and redistribution of flotation feed that xanthate and pine oil are brought into thorough contact with the pulp.

Flotation feed is separated into sand and slime products as previously stated. The slime product is not a true primary slime, but a combination of primary slime and gangue slime produced in the single pass primary rod mills. Table 3 gives the general results of separate flotation treatment for sands and slimes as compared to combined treatment of these products. The information contained in this table was obtained during a test period to determine the advantages of separate treatments.

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Table 3.—Separate sand and slime flotation compared with combined treatment

	Sand	Slime	Total sand and slime	Combined treatment
Analyses of feed:				
Total copper, per cent.....	2.48	1.60	2.03	2.04
Acid soluble copper, per cent.....	0.12	0.15	0.14	0.14
Iron, per cent.....	13.7	4.8	9.2	9.3
Analyses of concentrates:				
Total copper, per cent.....	11.30	13.54	12.07	12.13
Iron, per cent	37.6	30.4	35.1	36.2
Insoluble, per cent.....	4.8	15.1	8.3	6.4
Analyses of tailings:				
Total copper, per cent.....	0.18	0.22	0.20	0.22
Acid soluble copper, per cent.....	0.07	0.12	0.10	0.10
Recovery of copper, per cent	94.24	87.67	91.67	90.86
Ratio of concentration.....	4.83	9.65	6.49	6.54

A double metallurgical advantage for separate treatment of sand and slime may be noted in the foregoing tabulation. The first and most obvious is the lower total copper content in the tailings. The second advantage is that in the average concentrates analysis of sand and slime treatments, the iron is 1.0 per cent lower, equivalent to 2.0 per cent of pyrite content, and the insoluble is 2.0 per cent higher than the concentrates obtained by combined treatments. Thus the separate treatment replaces 2.0 per cent of pyrite with 2.0 per cent of insoluble thereby making a better fluxing material for smelting. Aside from these cited advantages new lines of attack may be launched against the two major metallurgical problems, recovery and ratio of concentration. Under the old method of combined sand and slime flotation, a drive for lower tailings was invariably circumscribed by ratio of concentration requirements and vice versa. With the separate treatment of sand and slime products the two problems are largely segregated into two distinct phases. Test work of this nature is now being conducted at this plant.

DEWATERING AND HANDLING OF CONCENTRATES

Figure 8 gives a flow sheet of dewatering and handling of concentrate. Finished concentrates from the flotation machines of each unit are fed to one 4-foot 6-inch by 18-foot DSC Dorr classifier which produces coarse concentrates and fine concentrates in order to facilitate dewatering.

The coarse concentrates from the rakes of the Dorr classifier, amounting to approximately 30 per cent of the total concentrates produced, are conveyed by launder to Blaisdell tanks located over railroad tracks at the side of the filter building. The Blaisdell tanks, four in number, are 30 feet in diameter and 12 feet deep and are constructed with cocoa-mat filter bottoms. These tanks are equipped with two distributors and one excavator for loading and unloading; they are operated in successive cycles consisting of filling, drying, and excavating and will dewater a full charge of 450 to 500 tons of coarse concen-

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trates in 15 to 18 hours, depending upon the character of the concentrates handled. A vacuum, amounting to 20 inches of water, is maintained throughout the dewatering period. The moisture contained in the coarser concentrates as delivered to the smelter amounted to 6.50 per cent during the year. During filling periods the overflow product from the Blaisdell tanks is conveyed by launder to a 24-foot Dorr thickener, the thickened pulp of which is fed to Oliver filters and the overflow product delivered, by Krogh pumps, to the fine concentrates pulp overflowing the concentrates classifier.

The fine concentrates pulp overflowing the Dorr classifier contains 25 per cent of solids and is conveyed by launder to one 75-foot Dorr thickener. This thickener overflows clear water and produces a thickened pulp containing approximately 60 per cent of solids which is conveyed by launders to bucket elevators for elevation to the Oliver filters. Four Oliver filters are installed but seldom more than one is used at one time. These filters are 14 by 14 feet in size and are equipped with oscillating agitators, steel tanks, steel arms and rims, and redwood staves. Canvas is used as filter cloth and is wound with No. 10 gage steel wire. Wire of this gage is selected because its life, amounting to approximately six months, is practically the same as the life of the canvas.

The filters are driven by 3-horsepower motors through worm gear speed reducers, chain, and sprocket. A 20 inch vacuum is maintained on the filters while the air for the blowing segment just above the scraper is at a pressure of 20 pounds. The filter cake produced during the year 1929 averaged 11.33 per cent of moisture and accounted for 70 per cent of the total concentrates produced.

DEWATERING AND DISPOSAL OF TAILINGS

A system of launders conveys the tailings from the flotation machines to one 200-foot traction thickener and two 75-foot auxillary thickeners. The per cent of solids in the underflow from the thickeners is approximately 50 per cent but varies somewhat according to the specific gravity and character of the solids handled. Tailings with high iron content require more water for conveyance to the tailings dams than material of lower specific gravity. The underflows from the thickeners are conveyed to the tailing dams by an unlined launder 23 inches wide and 10 inches deep. A general slope of 1.6 inches is maintained at the dams for all launders.

During the history of concentrator operations, six dams have been built, of which three are in present use. Gravity flow was used entirely until 1926 when it became necessary to elevate the tailings 60 feet to effect sufficient elevation on the outside dams. The dams in use at present were started with a toe thrown up by steam shovel. The launder trestle is built just inside of the toe, and tailings are deposited next to the toe with sublaunders or spouts, the water and slimes flowing toward the center of the dam. A cribbed tunnel and well in the center of the dam convey clear water to a pump house on the outside of the dam. Launder trestles are 25 feet in heighth for each bench elevation and are cross-braced with bents at intervals of 14 feet. A walkway is provided as an integral part of each launder and trestle structure and acts also as suspension support for the sublaunders.

Due to the high alkalinity content and character of the ore, no difficulty is experienced in recovering clear water, all of which is pumped back to the mill water system. It is estimated that 40 per cent of all reclaimed water is returned from the dams through one pumping station. The pump station employs two 2-inch centrifugal pumps in series and elevates the return water a total of 220 feet to the pump sump at the mill.

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After a building period varying from four to eight weeks, tailings are diverted. A certain section is dried and a border is thrown up by manual labor. This border insures against breakage by giving the outside of the dam a hard-packed and crusted surface. During the building of the border, all shrinkage cracks are tamped and any unlikely looking places are reinforced with obsolete cribbing and worn-out mill rods. A section of border two bents in length, or 28 feet, is considered a man-shift in this operation. In cross section the border is 4 feet wide at the base, 30 inches wide at the top, and 3 feet high. The outside continues the contour of the dam. Dusting at the dams is of no consequence because of lime incrustation and the relatively high iron content of the tailings.

During 1929 between 3,600 and 3,700 dry tons of tailings were handled daily.

ORE CONVEYING IN CRUSHING PLANT

Conveyors 1 and 2[4]

The conveying at this plant starts after the first coarse crushing operation. At this point two 54-inch pan type conveyors 61 feet long operating in parallel elevate the material 21 feet to accommodate the placing of 1,000-ton capacity surge bins between the first and second coarse crushing stages. These conveyors are driven through triple reduction gears and have a rated capacity of 1,800 tons per hour with a travelling speed of 100 feet per minute. The undersize product of the grizzly feeding the crusher lays a covering of fines several inches deep on these conveyors to absorb the wear and shock from the crusher discharge product containing ore lumps up to 12 inches maximum size. Repair costs on these conveyors have been materially reduced by the adoption of steel wheels and Alemite lubrication systems.

Conveyors 3 and 4

Four 48-inch apron feeders, 5 feet 6 inches between centers, draw ore from the surge bins and load two parallel 48-inch conveyor belts. The belts are 34 feet long, travel at the rate of 149 feet per minute, and elevate the ore a distance of 11 feet, discharging on the grizzlies which feed the gyratory crushers. These belts regulate the feeding rate to the gyratory and disk crushers and have a capacity of 300 tons per hour. Provision is made for starting and stopping by clutches.

Conveyors 5 and 6

The gyratory crusher discharge products of 4-inch maximum size, together with the grizzly undersize product which by-passes the gyratory crushers, are handled by two 36-inch conveyor belts 38 feet in length. These belts travel at a speed of 159 feet per minute, have a capacity of 300 tons per hour each, and elevate the ore a distance of 13 feet to feed the four Symons disk crushers.

Conveyor 7

One 42-inch conveyor belt, 64 feet in length, collects the Symons disk crusher products and the grizzly undersize products which by-pass the Symons machines. This belt travels at a speed of 254 feet per minute and has a capacity of 600 tons per hour. The belt is horizontal and discharges on No. 8 conveyor belt.

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4 - Conveyor numbers refer to numbers given in Figure 3.

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Conveyor 8

The main carrier belt from the crushing plant to the fine ore bins is 42 inches wide and 320 feet long. This belt travels at the rate of 350 feet per minute and effects an elevation of 89 feet with a capacity of 600 tons per hour. A Merrick weightometer on this belt provides a control on tonnage crushed.

Conveyor 9

An intermediate 42-inch conveyor 99 feet long carries the fine ore from the long incline conveyor No. 8 to the shuttle conveyor No. 9A over the fine ore bins. This intermediate belt runs at a speed of 329 feet per minute and has a rated capacity of 600 tons per hour. This conveyor elevates the ore a distance of 2.5 feet.

Conveyor 9A

The shuttle conveyor has a 42-inch belt, is 98 feet long, and travels at a speed of 335 feet per minute. It has a capacity of 600 tons per hour and is reversible in direction in order to distribute the crushed ore throughout the length of the fine ore bins.

ORE CONVEYING IN CONCENTRATOR

The belt conveyors are identical for the four concentrator units. The conveyors for one unit are as follows.

Conveyor 10[5]

Six apron feeders, 1 foot 8 inches by 7 feet 4 inches, deliver ore from the fine ore bins to a collector belt 24 inches wide and 46 feet long. This collector belt travels at the rate of 100 feet per minute and can handle 60 tons of ore per hour.

Conveyor 11

The collector belt, No. 10, discharges on a 24-inch cross belt, No. 11, which is 27 feet long. This belt carries the ore out from under the bins and will elevate 60 tons of ore per hour a distance of 10 feet to the main unit feeder belt.

Conveyor 12

The main unit feeder belt is 24 inches wide and 84 feet long. It travels at the rate of 150 feet per minute and elevates the ore a distance of 30 feet. When loaded it conveys 60 tons of ore per hour. Each of the four No. 12 belts is equipped with a Merrick weightometer for determining the unit tonnage. The weightometers on these No. 12 belts also serve as a check on the previously noted weightometer of No. 8 belt.

From this point on the pulp flows in launders by gravity to the secondary mills and the flotation cells. Each secondary mill is equipped with a two-way elevating scoop for elevating the ground product to the closed circuit classifier alongside each mill. All classifier overflow products are conveyed to the flotation feed distributors and thence to

5 - Conveyor numbers refer to numbers given in Figure 5.

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the flotation machines by 8 and 10 inch pipes because of insufficient fall for launder conveyance.

As previously noted, rougher concentrates are returned to the head of the flotation machine. This operation is done with two 6-inch Wilfley sand pumps. Flotation products are conveyed by launders to the dewaterers and the thickeners.

In the filter plant, fine concentrates are elevated with two bucket elevators, each elevator having 106 buckets 7 by 7 by 16 inches. These elevators travel at the rate of 433 feet per minute and elevate the concentrates pulp a distance of 65 feet to the Oliver filters. Coarse concentrates are elevated to the Blaisdell tanks, a distance of 12 feet, by air lifts.

Tailings from the thickeners are conveyed by launder to the dams, as previously described, and at the dams are elevated a distance of 60 feet by 6-inch Wilfley sand pumps.

SAMPLING

Sampling for Metallurgical Accounting

The first sample taken in the plant is that of the Symons crusher discharge product. This sample is used for a screen analysis in connection with the efficiency of grinding operations and also for the determination of moisture content.

All control assays are determined on pulps taken with automatic samplers. A centrally controlled system of automatic samplers cuts samples of feed, concentrates, and tailings pulps every 18 minutes. These samples are caught in buckets, and the completed samples represent 8-hour shifts for individual assays. The shift samples are in turn composited, on a tonnage basis, for daily and monthly analyses.

The sampling system is actuated by a balanced water tipple box. This box operates four-way valves on each down throw, thus relieving and building up the pressures on twin hydraulic pistons. An endless wire cable continues the throw of the hydraulic pistons and moves the individual sample cutters across the various pulp streams. Each 18-minute sample pulp flows through a splitter box equipped with two Jones type splitters, the final sample reaching the sample bucket. In this manner relatively large samples are cut from the pulp streams and quartered as cut, the weight of the composited 8-hour sample in each bucket being approximately 3,000 grams.

The concentrator feed or heads samples are taken after all crushing and grinding operations have been completed. As all products of grinding operations are brought to either sand or slime flotation distributor boxes, the samples of feed pulps are cut across the down spouts leading into the distributor boxes, one sample of sand flotation feed and one of slime flotation feed being taken.

A general feed sample is cut by hand from the primary rod mill discharge pulp. This sample is used primarily for control of grinding operations, but an analysis of the sample is also made for the purpose of checking the sand and slime flotation feed samples.

Samples of concentrates and tailings pulps are taken immediately after these products leave the flotation machines.

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The weighing of the ore and the checking of the weight by two sets of weightometers has been noted. The copper receipts to the concentrator are computed from the weight of ore with moisture contained, the moisture determination, and the copper analysis.

All sample pulps are collected every eight hours and taken to the screening and bucking room. The pulps are dewatered in a battery of vacuum filters rather than by decanting. A sketch of the filter used is given in Figure 9.

After the sample pulps are dewatered in the vacuum filters, they are taken just to dryness in an electric oven at a temperature of 400°F., care being taken not to burn or oxidize any samples. After pulverizing, a 24-hour composite sample is made up on a tonnage basis for assay and for a check on sampling operations.

The iodide method is used exclusively at the concentrator for copper analyses, the composite and feed samples being analyzed in duplicate. As a further check on analyses, monthly composites are analyzed at the concentrator and by the electrolytic method at the corporation laboratory at Douglas.

Ratio of concentration computed from the well-known formula, $\frac{C - T}{H - T}$, determines the dry weights of concentrates and tailings in tons. The indicated extraction is derived from the usual formula, $\frac{(H - T) \times C}{(C - T) \times H}$.

The final monthly settlement figure of copper produced in the concentrates is determined at the smelter. The accuracy of the sampling, assaying, and accounting system, as described, may be noted by the following tabulation showing the accounting of copper for the year 1929.

Indicated copper recovered at concentrator, pounds	Actual copper credited by smelter, pounds	Copper, unaccounted for, per cent
56,823,740	56,847,058	0.04

Sampling for Metallurgical Control

Colorimeter Determinations.- A system of close metallurgical control, by means of a colorimeter, is used because of considerable variation in copper content of concentrator feed. Under this system of control, the colorimeter operator furnishes the shift boss in charge of operation with hourly copper analyses of feed, concentrates, and tailings, the indicated extraction, and the ratio of concentration. The method employed consists in utilizing the rejects of the Jones splitters which, as previously noted, are used for cutting the samples of feed, concentrates and tailings taken by the mechanical samplers. These rejects are collected every hour, dewatered in a vacuum filter and dried on an electric hot plate. The sample for analysis is given a standard nitric acid and potassium chlorate digestion. Ammonia is added to precipitate the iron, and after filtering, the copper in the familiar blue copper ammonium hydrate solution is determined by comparison with a standard solution made up from a standard pulp of known copper content by the treatment just described.

The colorimeter consists of two graduated glass cylinders mounted over a reflected light with an eyepiece equipped with two mirrors, one superimposed above each cylinder. One

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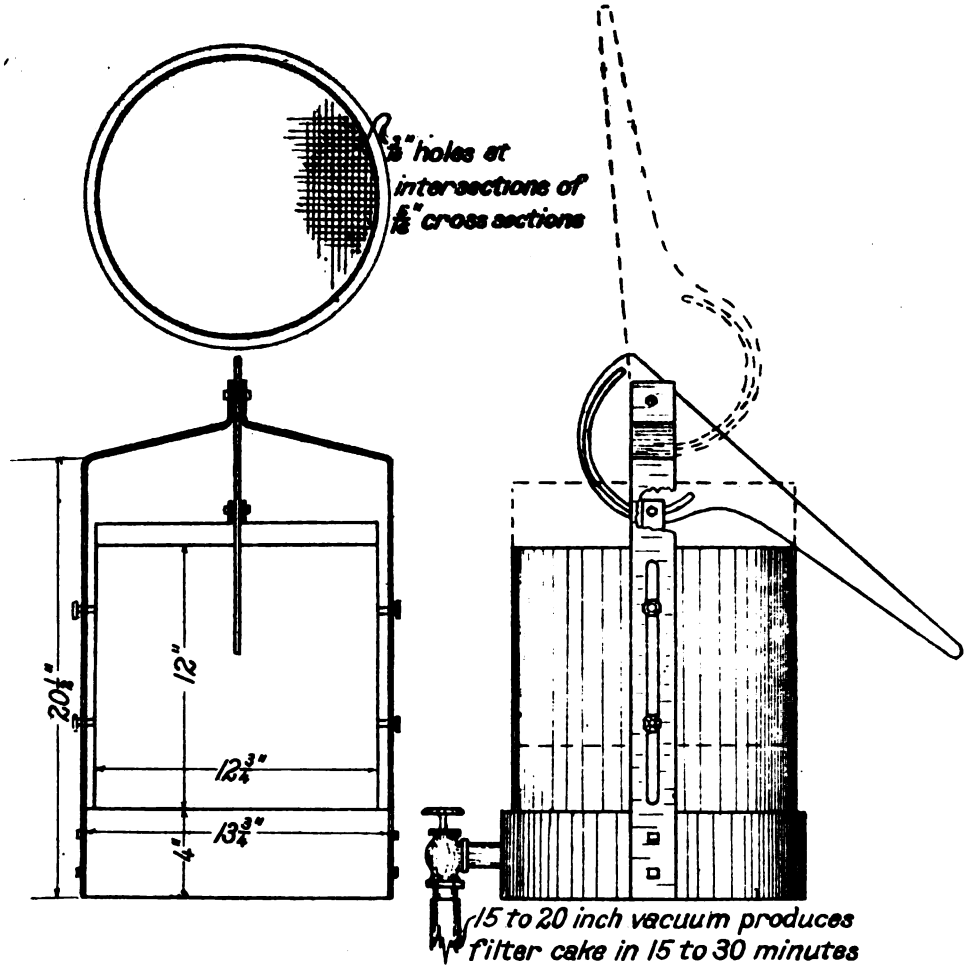
3/8" holes at intersections of 5/16" cross sections
20 1/2"
12"
12 3/4"
13 3/4"
4"
15 to 20 inch vacuum produces filter cake in 15 to 30 minutes

FIGURE 9.—Vacuum filter used for dewatering sample pulps

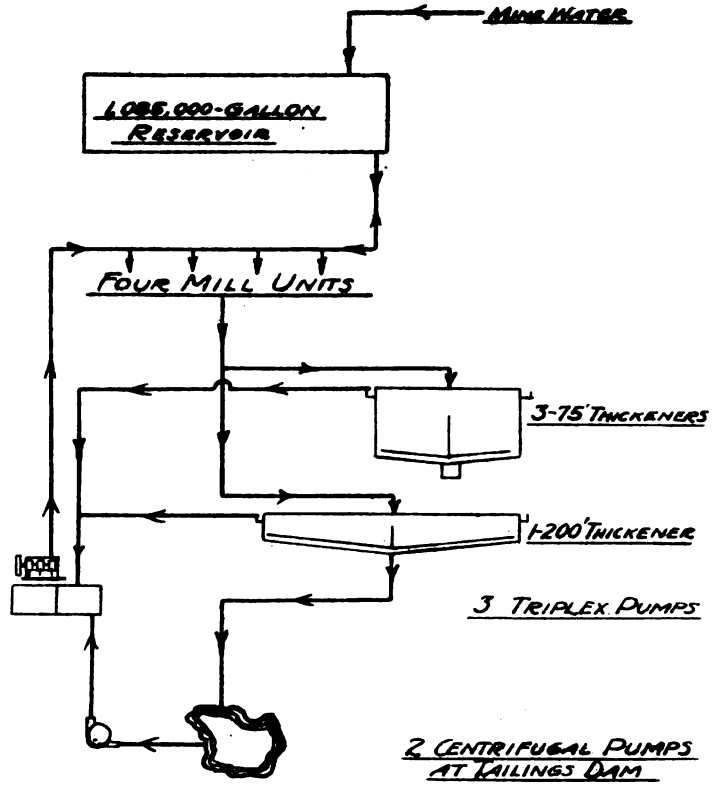

FIGURE 10.—Flow sheet of water recovery

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cylinder contains the standard solution and the other the solution being analyzed. The level of the solution in the standard cylinder is regulated by an outside tube and plunger. By manipulating this plunger, the level of the standard solution is raised or lowered, which changes the color density to permit direct comparison with the color of the unknown solution. When the two colors, reflected in the mirrors of the eyepiece, are matched, a reading is taken on the standard graduated cylinder. The apparatus requires little space and is conveniently located in the flotation department of the mill. As stated, samples are taken hourly and determinations are known within 30 minutes from the time of collecting the samples. Copper analyses so obtained check standard potassium iodide assays within 5.0 per cent of the total copper content.

Alkalinity Determinations

Samples of flotation feed are taken every hour and titrated for alkalinity. The method used has been discussed in detail under the heading of "Flotation reagents."

Sampling for Mechanical Efficiency

Hourly samples are taken by hand of the primary mill discharge product, the bowl classifier sand and overflow products, and the circulating classifier overflow product. These samples, together with certain previously noted automatic samples, are used as a check on mechanical efficiency. Screen analyses are made daily on composites of each of these samples, and an accurate determination of per cent of solids is also made. By this method a daily control is kept on the grinding efficiency of primary mills, secondary mills, and all classifying equipment. Assay screen analyses are made monthly and every third month's composite tailings sample is screened and given a complete microscopic analysis.

Hourly samples are also taken of all overflow products for immediate determination of solids content. The method of determining solids consists in weighing a known volume of pulp sample. Varying percentages of solids are standardized for varying ore conditions.

WATER SUPPLY

The cost of water at the Copper Queen concentrator is an appreciable item due to the necessity of pumping for a distance of 1,800 feet underground. This source of water was developed during a campaign for water to supply concentrator requirements; ample water was found, the only disadvantage being in the pumping cost. The actual pumping and delivery of water to the concentrator does not come under concentrator supervision. This is handled by the Copper Queen Branch mechanical department from which the concentrator buys the water.

The total water used per ton of ore concentrated amounts to between 1,400 and 1,500 gallons. Approximately 80.0 per cent of the water used is make-up water. The new water consumption per ton of ore milled during 1929 amounted to 275 gallons and the cost of new water amounted to $0.06 per ton of ore milled.

As this new water is from a deep source, it is very pure and carries no constituents harmful in mechanical or metallurgical operations. Figure 10 gives a flow sheet for recovering water. This subject has been described under the heading of "Tailings disposal."

Concentrator Recovery, Grade of Concentrates, and Tailings Losses

Table 4 gives a summary of copper recovery for the 1929 period.

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Table 4.—Summary of copper recovery, 1929

	Weight with moisture, tons	Moisture, per cent	Dry weight, tons	Analyses, per cent		Weight of copper, pounds	Per cent of total copper
				Total copper	Acid soluble copper		
Ore concentrated	1,492,664	3.16	1,445,425	2.15	0.13	62,189,430	100.00
Concentrates produced	244,190	9.91	219,991	12.92	0.31	56,823,740	91.37
Tailings	-	-	1,225,434	0.22	0.09	5,365,690	8.63

Because of the high pyritic iron content in the concentrator feed and the intimate association of the copper and iron minerals, a comparatively high-grade concentrate is as yet unattainable. The following chemical analysis will show the main diluent factors affecting grade of concentrates.

Analysis of concentrator feed

Per cent								
Total copper	Iron	Sulphur	SiO_2	Al_2O_3	MgO	CaO	Res.	Total
12.92	35.0	43.1	5.7	1.7	0.1	0.6	0.88	100.0

In common with other similar metallurgical plants, an economic balance is maintained in respect to grade of concentrates and copper loss in tailings. This balance takes into consideration not only the metallurgy of the concentrator, but also the metallurgy of the smelter.

The degree of fine grinding is a direct function of the grade of concentrates at this plant. It has been demonstrated that grinding of the flotation feed to 55.0 per cent through 200-mesh size is sufficient to effect a satisfactory recovery, but it is necessary to further grind flotation feed between 70.0 and 75.0 per cent through 200-mesh size to bring about the maximum in grade of concentrates. The set standard maintained at this plant of grinding 75.0 per cent through 200-mesh size approximates very closely the economic grinding limit as determined by the derived benefits.

Following are assay-screen analyses of concentrates and tailings for a recent month, segregated into concentrates and tailings from sand and slime flotation.

Assay screen analyses of concentrates and tailings

CONCENTRATES FROM SAND FLOTATION					
Screen size, mesh	Weight, per cent	Analyses, per cent		Per cent of total	
		Total copper	Iron	Copper	Iron
On 100	0.4	3.74	25.1	0.12	0.27
On 150	4.1	5.35	33.6	1.79	3.73
On 200	10.1	6.10	39.5	5.04	10.82
Through 200	85.4	13.34	36.8	93.05	85.18
Composite	100.0	12.24	36.9	100.00	100.00

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Assay screen analyses of concentrates and tailings - Continued

CONCENTRATES FROM SLIME FLOTATION

Screen size, mesh	Weight, per cent	Analyses, per cent		Per cent of total	
		Total copper	Iron	Copper	Iron
On 100	3.9	5.85	32.9	1.69	3.95
On 150	9.2	7.82	35.9	5.33	10.16
On 200	10.3	9.00	36.8	6.86	11.66
Through 200	76.6	15.19	31.5	86.12	74.23
Composite	100.0	13.51	32.5	100 00	100.00

TAILINGS FROM SAND FLOTATION

Screen size, mesh	Weight, per cent	Analyses, per cent		Per cent of total	
		Total copper	Acid soluble copper	Copper	Acid soluble copper
On 65	0.6	0.31	0.05	1.02	0.37
On 100	7.6	0.19	0.04	7.94	3.73
On 150	13.4	0.16	0.04	11.78	6.58
On 200	9.4	0.14	0.04	7.23	4.62
Through 200	69.0	0.19	0.10	72.03	84.70
Composite	100.0	0.18	0.08	100.00	100.00

TAILINGS FROM SLIME FLOTATION

On 65	2.4	0.37	0.03	3.34	0.43
On 100	6.7	0.31	0.04	7.82	1.60
On 150	9.2	0.23	0.04	7.97	2.27
On 200	7.4	0.19	0.05	5.30	2.22
Through 200	74.3	0.27	0.21	75.57	93.48
Composite	100.0	0.26	0.16	100.00	100.00

A microscopic analysis of the concentrates is not available at this writing. The feature of such a study, however, will show a heavy chalcocite-pyrite association, especially in the plus 200-mesh sizes. A general summary of results of the most recent microscope examination of tailings products follows.

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Distribution and associations of copper minerals in sand flotation tailings

	Copper, content per cent		Per cent of total					
			Sulphide copper		Oxide copper		Total copper	
Sulphide copper:								
Chalcocite, free	0.09209	-	80.8	-	-	-	49.7	-
Chalcocite, associated with gangue	0.01827	-	16.1	-	-	-	9.9	-
Chalcocite, associated with pyrite	0.00192	-	1.7	-	-	-	1.0	-
Chalcocite, total	-	0.11228	-	98.6	-	-	-	60.6
Chalcopyrite	0.00085	-	0.7	-	-	-	0.45	-
Covellite	0.00085	-	0.7	-	-	-	0.45	-
Total covellite and chalcopyrite	-	0.00170	-	1.4	-	-	-	0.9
Total sulphide copper	-	0.11398	-	100.0	-	-	-	61.5
Oxide copper:								
Cuprite, 65 per cent soluble	0.03152	-	-	-	44.2	-	17.0	-
Other acid soluble copper minerals	0.03982	-	-	-	55.8	-	21.5	-
Total oxide copper minerals	-	0.07134	-	-	-	100.0	-	38.5
Total copper minerals	-	0.18532	-	-	-	-	-	100.0

Distribution and association of copper minerals in slime flotation tailings

	Copper content, per cent		Per cent of total					
			Sulphide copper		Oxide copper		Total copper	
Sulphide copper:								
Chalcocite, free	0.05764	-	61.1	-	-	-	22.9	-
Chalcocite, associated with gangue	0.02298	-	24.3	-	-	-	9.1	-
Chalcocite, associated with pyrite	0.00413	-	4.4	-	-	-	1.6	-
Chalcocite, total	-	0.08475	-	89.8	-	-	-	33.6
Chalcopyrite	0.00938	-	9.9	-	-	-	3.7	-
Covellite	0.00025	-	0.3	-	-	-	0.1	-
Total chalcopyrite and covellite	-	0.00963	-	10.2	-	-	-	3.8
Total sulphide copper	-	0.09438	-	100.0	-	-	-	37.4
Oxide copper:								
Cuprite, 65 per cent soluble	0.04923	-	-	-	68.8	-	19.5	-
Other acid soluble copper minerals	0.10869	-	-	-	31.2	-	43.1	-
Total oxide copper minerals	-	0.15792	-	-	-	100.0	-	62.6
Total copper minerals	-	0.25230	-	-	-	-	-	100.0

It is interesting to note the ratio of oxide to sulphide copper in the slime tailings analysis. The oxide minerals are chiefly cuprite and chalcanthite with other basic copper oxides which are acid soluble but not water soluble. Over 90 per cent of the oxide copper occurs in the minus 200-mesh size, and the microscopist records these copper oxide minerals appearing as dust and surface alteration on the chalcocite.

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Metallurgical Data

Table 5 gives screen analyses of concentrator products for December, 1929. The metallurgical data sheet for 1929 is given in Table 6, and the metallurgical balance sheet for 1929 in Table 7. Table 8 shows distribution of power for 1929.

Table 5.--Typical screen analyses as determined in December, 1929

Size	Weight, per cent								
	Run of mine	Jaw crusher discharge	Gyratory crusher discharge	Symons crusher discharge	Primary rodmill discharge	Bowl classifier overflow	Bowl classifier sand	Circulating classifier overflow	Flotation feed
On 1.75 inch	(1)	(2)	(3)	1.2	-	-	-	-	-
On 1.50 inch	-	-	-	1.5	-	-	-	-	-
On 1.25 inch	-	-	-	4.4	-	-	-	-	-
On 1.05 inch	-	-	-	12.7	-	-	-	-	-
On 0.743 inch	-	-	-	17.0	-	-	-	-	-
On 0.525 inch	-	-	-	11.4	-	-	-	-	-
On 3 mesh	-	-	-	14.4	-	-	-	-	-
On 4 mesh	-	-	-	4.9	-	-	-	-	-
On 6 mesh	-	-	-	3.4	-	-	-	-	-
On 8 mesh	-	-	-	2.4	-	-	-	-	-
On 10 mesh	-	-	-	2.5	-	-	-	-	-
On 14 mesh	-	-	-	1.4	-	-	-	-	-
On 20 mesh	-	-	-	1.4	0.9	-	2.0	-	-
On 28 mesh	-	-	-	1.6	2.9	-	5.5	-	-
On 35 mesh	-	-	-	1.3	5.3	-	11.2	-	-
On 48 mesh	-	-	-	1.3	8.8	-	18.2	-	-
On 65 mesh	-	-	-	1.1	9.6	2.4	18.3	1.0	1.7
On 100 mesh	-	-	-	1.3	13.0	5.9	18.4	5.1	5.5
On 150 mesh	-	-	-	0.8	8.1	7.5	8.7	8.7	8.1
On 200 mesh	-	-	-	0.6	5.8	8.3	4.9	11.7	10.0
Through 200 mesh	-	-	-	13.4	45.6	75.9	12.8	73.5	74.7
Total	-	-	-	100.0	100.0	100.0	100.0	100.0	100.0

(1) From mine grizzlies, 20-inch maximum size.

(2) Unchanged from previous steam shovel operations, 12-inch maximum size.

(3) Four-inch maximum size.

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Table 6.--Metallurgical data for year 1929

Total ore treated, dry tons	1,445,425
Days operated	337.61
Hours operated per day	24
Average ore treated per 24 hours, tons	4,281
Average sections running	4
Average ore treated per 24 hours per section, tons	1,070
Total concentrates produced, dry tons	219,991
Average concentrates produced per 24 hours, dry tons	652
Analysis of concentrates:	
Copper, per cent	12.92
Insoluble, per cent	7.4
Moisture, per cent	9.91
Copper produced in concentrates, pounds	28,411,870
Recovery of total copper by analysis, per cent	91.32
Recovery of sulphide copper, per cent	94.58
Ratio of concentration, tons into one	6.58
New water consumption per ton of ore, gallons	275
Power consumed per ton of ore milled, kilowatt hours	13.70
Total rod and ball consumption per ton of ore milled, pounds	2.75
Total liner consumption per ton of ore milled, pounds	0.53
Total steel consumption per ton of ore milled, pounds	3.28
Reagents used per ton of ore milled, pounds:	
Potassium xanthate	0.12
Pine oil	0.042
Lime	4.915

Table 7.--Metallurgical balance for year 1929

	Weight, dry tons	Analyses, per cent							
		Copper		Silica	Iron	Sulphur	Alumina	Magnesia	Lime
		Total	Acid soluble						
Mill feed	1,445,425	2.15	0.13	61.6	8.9	10.1	10.9	0.1	0.9
Concentrates	219,991	12.92	0.31	5.7	35.0	43.1	1.7	0.1	0.6
Tailings	1,225,434	0.22	0.09	71.6	4.2	4.1	12.6	0.1	1.0

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Table 8.—Power data for year 1929

	Kilowatt hours per ton	Per cent of total
Crushing and conveying:		
Jaw crusher	0.12	0.92
Gyratory crushers	0.13	0.95
Disc crushers	0.30	2.16
Conveying	0.46	3.31
Lighting	0.02	0.13
Cranes	—	0.02
Dust collector system	0.02	0.14
Total crushing and conveying	1.05	7.63
Concentrator and tailings dam:		
Primary grinding	4.73	34.54
Secondary grinding	4.05	29.59
Classifiers	0.23	1.68
Cranes	0.01	0.06
Concentrates classifiers	0.01	0.09
Flotation	1.66	12.11
Concentrates handling	0.42	3.03
Mill water	1.22	8.92
Mill shops	0.01	0.15
General expense	0.05	0.40
Lighting	0.16	1.14
Assaying and sampling	0.09	0.65
Experimental work	0.01	0.06
Total concentrator and tailings dam	12.65	92.37
Grand total	13.70	100.00

LABOR

An organization flow sheet of the concentrator personnel is presented in Figure 11.

This organization, as segregated according to working groups, accounted for the following average number of men per day employed during the 1929 period.

Average number of men employed per day, 1929

Crushing	Operation	Repairs	Tailings	Miscellaneous	Total
30.5	54.9	21.4	10.5	12.4	129.7

On a tonnage basis, operating labor produced 37.36 tons per man-shift and repair labor 189.56 tons per man-shift, giving a total production of 31.22 tons per man-shift for 1929.

As indicated, most of the construction work, all repair and machine work, and sundry similar work is executed by mill crews. Electrical work is under the supervision of the chief electrician for the Copper Queen Branch. No hiring of men is done at the concentrator proper; all applicants must go through the Copper Queen Branch employment department

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and are requisitioned by the mill as required. Timekeeping is also handled from a centralized Copper Queen Branch department. The concentrator maintains its own supply department, which is subsidiary to a main supply department.

All operating and mechanical work is done by Americans; Mexicans are employed as helpers, oilers, swampers, and clean-up men. One or two exceptions may be cited in the case of especially trained Mexican trademen.

SUMMARY OF COSTS

Table 9 gives a summary of costs per ton of ore milled for the year 1929.

Table 9.--Summary of costs per ton
Year 1929
Tons of ore treated, 1,445,425
Tons of concentrates produced, 219,991

	Operating labor and salaries	Power	Supplies	Reagents	Miscellaneous	Total
Crushing and conveying	$0.036	$0.008	$0.015	-	-	$0.059
Primary grinding	0.017	0.039	0.049	-	$0.006	0.111
Secondary grinding	0.012	0.035	0.036	-	-	0.083
Concentrating, flotation	0.010	0.014	0.004	$0.042	0.005	0.075
Concentrates handling	0.013	0.003	0.003	-	-	0.019
Assaying and sampling	0.011	0.001	0.002	-	-	0.014
Tailings disposal	0.012	0.006	0.004	-	-	0.022
Water supply	0.007	0.010	0.003	-	0.039	0.059
Miscellaneous	0.033	0.005	0.003	-	0.010	0.051
Total direct	$0.151	$0.121	$0.119	$0.042	$0.060	$0.493

A tabulation appearing under the heading of "History of concentrator operation" shows the continual reduction in cost per ton of ore milled since the start of operation in 1923. The following tabulation of costs by elements is abstracted from the costs for the year 1929.

Cost by elements, 1929

Element	Cost per dry ton milled	Per cent of total cost
Labor and salaries	$0.155	31.44
Total supplies	0.162	32.86
Power	0.120	24.34
Water and heat	0.043	8.72
Miscellaneous	0.018	3.65
Gross	0.498	101.01
Credit, smelting ore crushed	0.005	1.01
Total direct	$0.493	100.00

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The only item included in the foregoing tabulation which has resisted lowering from previous yearly figures is the water cost as described in previous comment.

A recapitulation of costs with respect to processes follows.

Costs by processes

Process	Cost per dry ton milled	Per cent of total cost
Crushing and conveying	$0.059	11.97
Primary grinding	0.111	22.52
Secondary grinding	0.083	16.84
Flotation	0.075	15.21
Concentrate handling	0.019	3.85
Tailings disposal	0.022	4.46
Assaying and sampling	0.014	2.84
Mill water	0.059	11.97
Mill department	0.051	10.34
Total direct	$0.493	100.00

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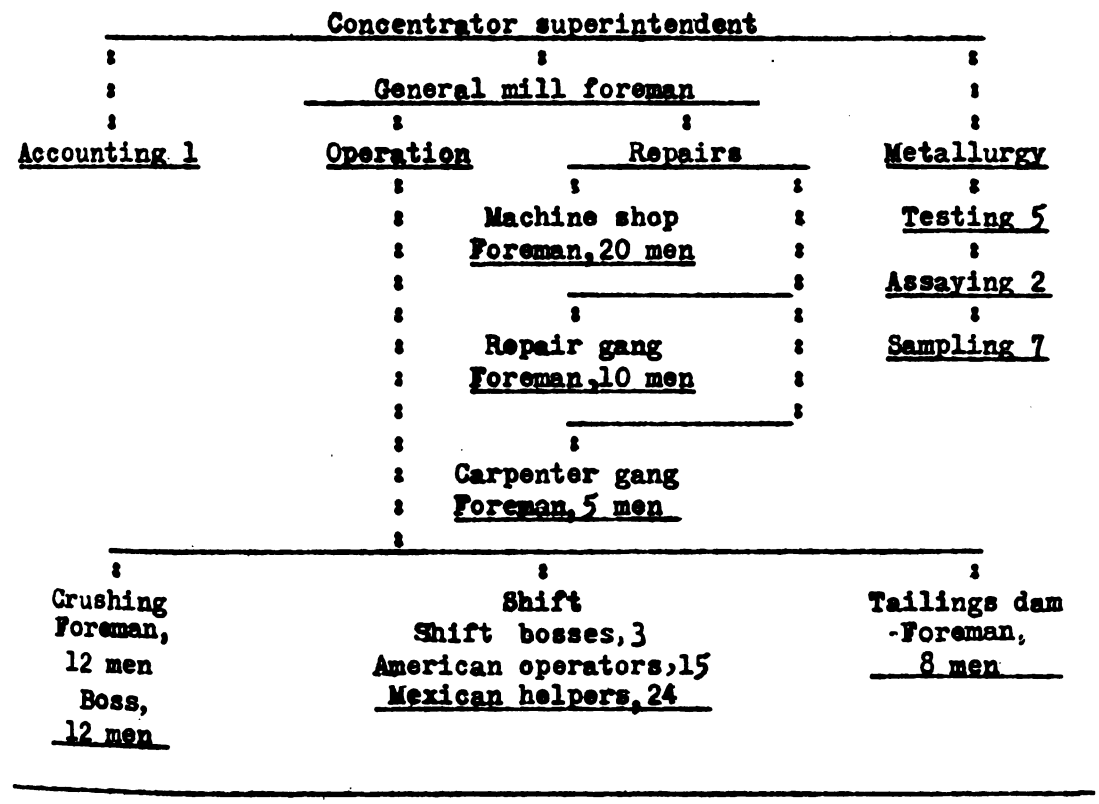

Figure 11.- Organization flow sheet

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